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MINING METHODS AND COSTS AT THE
BLACK BUTTE QUICKSILVER MINE,
LANE COUNTY, OREGON



BY

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MINING METHOD AND COSTS AT THE BLACK BUTTE QUICKSILVER MINE, LANE COUNTY, ORE.¹

By William W. Elmer²

INTRODUCTION

This paper, describing the mining method at the Black Butte quicksilver mine, is one of a series of papers being prepared by the U. S. Bureau of Mines on mining practices, methods, and costs in the various districts of the United States. It is presented to show the mining practice and costs in the successful treatment of low-grade quicksilver deposits. The operation discussed is an excellent example of what can be done in working a limited tonnage of low-grade ore under sound management.

The Black Butte mine is situated at Black Butte, Lane County, in the southern part of the State of Oregon, 17-1/2 miles from Cottage Grove -- a station on the Siskiyou branch of the Southern Pacific Co. The Black Butte mining district lies in the foothills of the Calapooia Mountain Range, which connects with the Cascade Range on the east and with the Coast Range on the west. The elevation at the reduction plant is 1,000 feet above sea level. The apex of Black Butte Mountain, on which the mine is situated, has an elevation of 2,700 feet. The district possesses exceptional natural facilities infrequently met with elsewhere for the mining and reduction of cinnabar ores. Several factors which contribute to the unusually low mining costs are:

(1) The climate is temperate, so that operations are carried on without difficulty the year around.

(2) Timber is abundant in the immediate vicinity of the mine.

(3) There is ample water for all purposes in the several creeks on the property.

(4) The ore deposits are easily workable, and the ground stands well without support.

(5) Shaft sinking and pumping are unnecessary, as the steep slope of Black Butte Mountain, in conjunction with the dip of the vein, affords excellent opportunities for exploitation of the ore deposits by means of adits at any level to cut the vein for a vertical distance of at least 1,000 feet below the apex of the mountain.

1 - The Bureau of Mines will welcome reprinting of this paper provided the following footnote acknowledgment is used:

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2 - Consulting engineer, U. S. Bureau of Mines.

(6) The topography also allows use of an easy method for gravity tram transportation from the present lowest mine level to the reduction plant.

(7) Hydroelectric power is obtainable at reasonable cost.

(8) The near-by farming lands are fertile and easily irrigated and supply the truck-garden products necessary for the camp.

(9) The labor supply from the surrounding communities is ample.

The mine and reduction plant employ a crew of about 40 men. At present about 140 tons of ore containing 3-1/2 to 4-1/2 pounds of mercury per ton are treated per day.

The author acknowledges the assistance of William O. Vanderburg, associate mining engineer, U. S. Bureau of Mines, in preparing this paper.

HISTORY

The principal vein in the district was discovered in the early nineties. Prior to 1900 only a small amount of development work was done on the property, and the equipment for the reduction of the ores consisted of a 40-ton Scott-Hutner furnace with series chamber condensers operated by natural draft. In 1898 the property passed into the hands of Mr. W. B. Dennis who, in succeeding years up to 1908, extended the workings of the former holders and opened new levels. A total of 15,000 feet of development work was done. The property lay idle until 1916 when, due to the high price of quicksilver, it was operated by a substantial mining organization under lease and bond until April, 1919. During this active period the Scott-Hutner furnace was rehabilitated and the natural draft altered to an artificial down draft furnished by two 42-inch suction fans, increasing the capacity about 25 per cent. An improved dust chamber, an intermediate water-cooled cast-iron pipe condenser, an elaborate supplementary condenser constructed of brick and concrete, and 18-inch clay pipes, water-sprayed, were also added.

Due to the fall in the price of quicksilver at the close of the World War, the property was closed down in 1919 and remained idle until 1927. Operations were resumed in the fall of 1927 under the management and ownership of the Quicksilver Syndicate. The old Scott-Hutner furnace has been dismantled and replaced by two rotary furnaces, each 60 feet in length and 4 feet outside diameter, having a normal combined capacity of 150 tons per 24 hours.

PRODUCTION

No records of production before 1916 are available. The subsequent production of quicksilver from the property has been as follows:

<u>Period</u>	<u>Number of flasks</u>
1916-1919.....	1,271
1927.....	139
1928.....	1,000
1929.....	<u>1,313</u>
Total 1916-1929.	3,723

GEOLOGY

The rocks in the vicinity of Black Butte, as exposed on the surface and in the underground workings, are all of igneous origin and consist of a series of andesitic rocks and lava tuffs intruded in places by basalt dikes. The rocks have been extensively altered by hydrothermal action.

The principal vein lies along and for more than 1,000 feet forms the apex of Black Butte Mountain. The vein outcrops for a total distance of 3,500 feet and at intervals rises boldly to heights of 30 to 50 feet above the northerly dipping slope of the mountain. The general strike of the main fissure is south 70° east, and the average dip is 58° to the northeast.

The ore occurs in fractures and as irregular lenses of low-grade material sometimes many feet in thickness where the mineralizing solutions have impregnated the porous country rock.

In general, the cinnabar occurs in very finely disseminated particles in the country rock near the main fissure and in seams tributary to the main fissure.

The width of the ore mined ranges from 15 to 70 feet. The ore deposits are well adapted to the shrinkage-stope system of mining, as the ground stands indefinitely without support, and the dip of the vein is such that the ore flows readily to the chutes by gravity.

PROSPECTING AND EXPLORATION

Prospecting in the vicinity of the Black Butte district is difficult, as the slopes of the surrounding mountains are covered with a heavy growth of timber and deep soil. Prospecting has been largely confined to surface trenches dug on the outcrop and adits driven to intersect the main vein at suitable depths. After the main fissure has been cut with an adit, drifts are extended along the strike of the fissure. Crosscuts are driven at intervals of about 50 feet, from the drifts into both hanging and foot walls. Ore shoots have been discovered by the crosscuts that would otherwise have been missed. In general the walls of the ore shoots are economic rather than structural.

SAMPLING AND ESTIMATING

In mining operations the grade of the ore is determined by visual inspection supplemented by panning, a method which, in the hands of an experienced manipulator familiar

with the ore, furnishes an estimate of the value per ton sufficiently close for mining purposes. For furnace control a 100-pound daily sample is taken at the reduction plant; this sample consists of "grabs" taken from the buckets of the gravity tram at the lower terminal; the ore is crushed to minus 1-1/2 inch at the mine. The 100-pound sample is dried and crushed to minus 60 mesh and 2 grams of the pulp are taken for each assay; the assays are run in duplicate. Daily sampling of burned ore is conducted in the same manner. The Whitton method has been found the cheapest and quickest method for assaying mercury ores at Black Butte. In this method as employed there, each of the 2-gram samples is combined with an amount of clean iron filings that slightly exceeds the weight of the pulp sample. The pulp and iron filings are placed in a 4-gram-capacity iron crucible, on top of which is placed a clean silver-foil cover. The silver foil is tightly sealed to the crucible, so that the mercury fumes can not escape. This is done by means of a small brass water pan and clamp, as shown in Figure 1. The crucible is heated for a period of 17 minutes with an ordinary gasoline torch with an upward-projecting flame. The mercury in the ore sample is volatilized and condenses upon and forms an amalgam with the silver foil. The difference in weight of the clean silver foil and the foil after amalgamation gives the weight of mercury in the pulp sample. After weighing the assay the silver foil is cleaned by burning in an alcohol flame. It is necessary to dry and weigh the silver foil carefully both before and after retorting.

For greater accuracy the author has used a gold plate weighing 1/2 ounce instead of the silver foil. By using the gold plate there is less opportunity for inaccurate weighing, as the plate can be cleaned more readily with dilute nitric acid. Another advantage in the use of the gold plate is that traces of amalgam can be visually detected more readily. This process is not precise, but with sufficient checking of assays (run always in duplicate in the writer's practice) it affords a dependable record.

DEVELOPMENT

The mine is developed by six main adit levels, as shown in Figure 3. The adits and drifts are untimbered and average about 6 by 7 feet in cross section.

An adit or drift is driven by a crew of three men, one miner and two muckers, on each of the two shifts. All work is done on company account. The ground drills readily and in most instances unmounted machines of the jack-hammer type are used. The 12-hole drift round is shown in Figure 2-a. The average depth of the rounds is about 5 feet.

All drill steel is cut in 2, 4, and 6 foot lengths and sharpened by hand. The usual cross bit is used with diameters of 1-1/4, 1-5/8, and 1-3/4 inches for the three changes. Drill-steel consumption is a very small item in the mining costs.

In blasting, 35 per cent strength gelatin dynamite is used with No. 6 detonators and fuse.

Drilling and shoveling are done simultaneously in advancing an adit or drift. The muckers also tram the loaded cars by hand to the nearest gathering switch, whence they are hauled by a horse to the portal in trains of six cars.

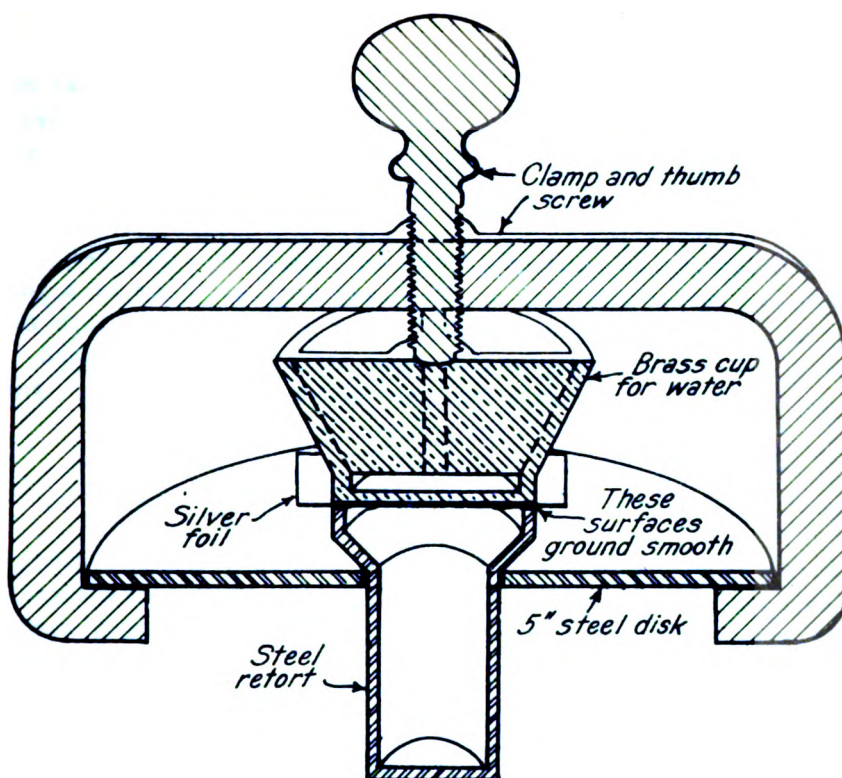


Figure 1.- Whitton mercury-determination apparatus

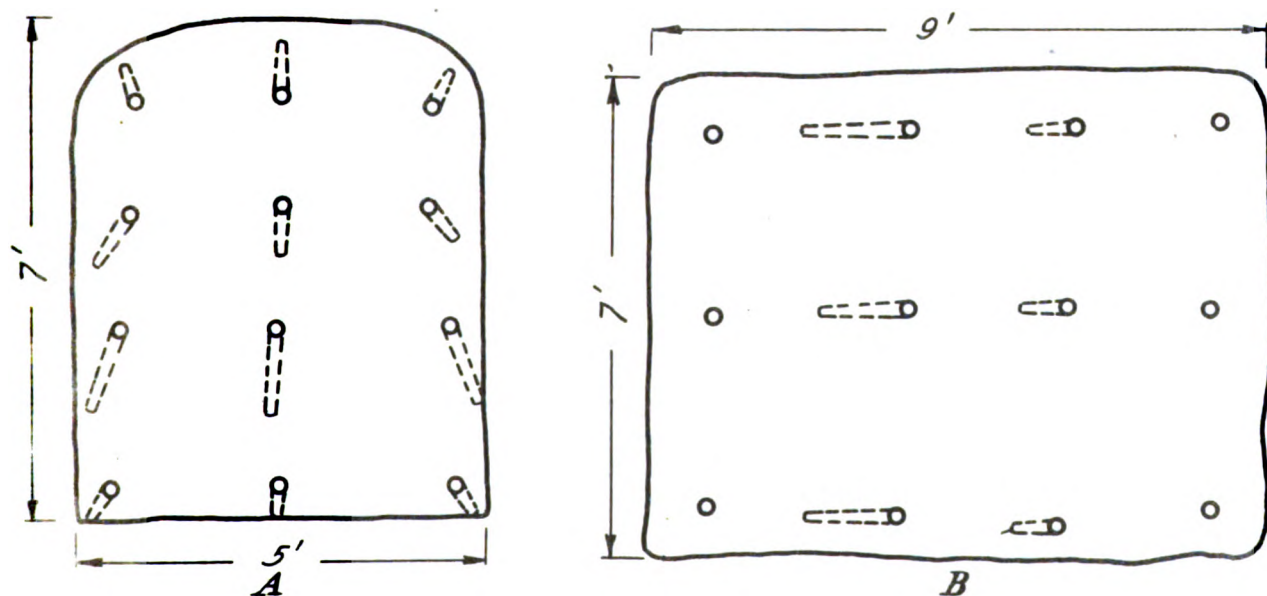


Figure 2.- Drift and raise rounds

Only one raise is driven to connect consecutive levels. Each raise is driven 7 by 9 feet in cross section and divided into chute and manway compartments. Stulls 10 to 12 inches in diameter placed on 6-foot centers are used for timbering. The chute compartment is lined with 2-inch plank lagging at the side adjacent the manway.

A crew of two men, one miner and one timberman, is employed on each of the two shifts in advancing a raise. The timberman cuts hitches and places the stulls in position while the miner is drilling. A raise round consists of 12 holes drilled to a depth of 5 feet (fig. 2-b). Stoper machines of the hand-rotated type, using 1-inch cruciform steel, are used in raising. Gelatin powder of 35 per cent strength is used and detonated with No. 6 caps and fuse.

STOPING

The ore body and the inclosing walls are both structurally strong. The ore is mined by the shrinkage stope method without artificial support being necessary. However, as stoping progresses upward occasional pillars are left behind because they are too low grade to mine economically.

The ore now being produced (January, 1930) comes from stopes above the 500, 600, and 900 foot levels. On the 500 and 600 levels the ore is hand trammed to the main ore pass raise and dropped through this raise to the 900 level (see map, fig. 2).

The average dimensions of the stopes are 20 feet wide, 300 feet long, and 200 feet on the dip.

In preparing a stope for mining raises are driven at intervals of 25 feet from the back of the drift to a height of 7 feet. These raises are belled out at their tops to connect with each other. In each raise a simple-type chute is built with 2-inch planks nailed to round posts and stulls. Chute gates consist of 2-inch planks sliding in wooden cleats nailed to the sides of the chute lips. Chute openings are 3 feet long by 2-1/2 feet wide.

When the chutes are installed and the belling-out process is completed stoping of the ore is begun. The stope face is carried upward horizontally. Sufficient ore is drawn off from time to time so that the distance from the top of the broken ore to the stope face is convenient for drilling and for barring down the loose ore. If the ore shoot gives out in carrying the stope upward prospect raises are driven and subdrifts run when convenient.

To afford access to the stopes and ventilate the workings manway raises are carried up as the face of the stope advances. These manways are constructed of round stulls and comprise one compartment 4 by 4 feet in the clear. Three manways are carried up for each stope. The sides of the manways adjacent the broken ore are lined with split lagging.

Supplies are carried into the stopes by hand.

No standard round is used in stoping operations. Holes and rounds are drilled under the supervision of the mine foreman, and by taking advantage of planes of weakness

yield a higher tonnage per man, as shown in Table 2. Drilling is done dry. Drilling and blasting are done on both shifts. In general, the ore breaks fairly fine; the occasional large boulders encountered are broken in the stops by plugging with jackhammers and blasting, or by breaking with hammers by hand.

Machines of the jackhammer type are used for drilling horizontal slabbing holes and hand-rotated stopers for drilling upward inclined to vertical holes. One miner will drill 9 to 15 holes per 8-hour shift. Average depth of holes is about 5-1/2 feet. Blasting is done with 35 per cent strength gelatin dynamite with No. 6 detonators and fuse.

TRANSPORTATION

The ore is trammed from the stope chutes to the portal of the 900 level adit in 1-ton, roller-bearing end-dump cars. Trimming is done by horse-drawn trains of six cars. The driver loads and dumps the cars. The average tramping distance is about 800 feet. Track consists of 16-pound rails laid to 18-inch gage on a grade of 1/2 per cent.

The ore is dumped from the cars at the portal of the 900 level adit and passes by gravity through a 15-inch jaw crusher set to crush to minus 1-1/2 inch. From the crusher the ore discharges into an inclined-bottom wooden bin, whence it is loaded into the buckets of a single-cable serial gravity tramway and lowered to the reduction plant, about 600 feet below the portal.

The tramway consists of 5,152 feet of 7/8-inch steel rope carrying 112 buckets, capacity 75 pounds each, spaced at intervals of 46 feet. The cable revolves about brake-controlled sheaves at the terminals. The tripod towers for supporting the cable are constructed of round timbers about 10 inches in diameter. The capacity of the tram is about 140 tons per 24 hours.

WAGE SCALE

The wage scale, as of December, 1929, for the principal classifications of employees was as follows:

Blacksmith.....	\$5.00
Blacksmith helper.....	4.50
Gravity tram operator..	4.25
Crusherman.....	4.00
Furnace operator.....	4.50
Miner.....	4.50
Mucker.....	4.00
Trammer (horse driver)	4.50
Timberman.....	4.50

All work is done on company account.

VENTILATION, FIRE HAZARD, AND GENERAL SAFETY CONDITIONS

The mine is well ventilated by natural ventilation, so that the working places are cool and comfortable the year around.

The amount of timber used in the shrinkage-stope method of mining as employed at Black Butte is very small. The small amount of timber underground and the excellent ventilation reduce the fire hazard to a minimum.

In advancing a shrinkage stope an effort is made to keep the back of the stope within 6 feet of the broken ore so that the stope can be readily inspected for loose slabs of rock which might fall and injure the workmen. No fatal accident has occurred in mining operations within the last three years.

Table 1. - Costs per ton, Black Butte mine
Jan. 1 to July 30, 1929

Tons of ore treated: 17,721

Mining method: Shrinkage.

	Labor	Superin- tendence	Power	Explosives	Oil	Other supplies	Total
¹ Mining	\$0.95	\$0.07	\$0.07	\$0.15	-	\$0.07	\$1.31
Treating	.38	.03	.07	-	\$0.57	.08	1.13
Total	1.33	.10	.14	.15	.57	.15	2.44

¹Development costs are included. Surface costs were not included. For comparison note that the surface costs for Nov., 1929, given below are 15 cents per ton.

Costs for November, 1929 - the last month for which records were available at the time this paper was written - are shown below:

Tons of ore treated: 3,626

Month of November, 1929

	Labor	Superin- tendence	Power	Explosives	Oil	Other supplies	Total
Mining	\$0.839	\$0.099	\$0.085	\$0.185	-	\$0.055	1.263
Treating	.272	.069	.048	-	\$0.692	.048	1.129
Surface	.097	-	.006	-	-	.047	.150
Total	1.208	.168	.139	.185	.692	.150	2.542

The costs as given in the above table are based on the tons of ore treated. An additional tonnage of ore was broken during the above periods. This tonnage remained in storage in the stopes.

Table 2. - Costs of mining and milling in units of labor, power and supplies

Black Butte mine		Month of November, 1929	
<u>Tons of ore treated: 3,626</u>		<u>Mining method:</u>	<u>Shrinkage</u>
A Labor (man hours per ton)			
Mining, including development			
Miners	0.402		
Muckers	.397		
Trammers	.253		
Timbermen	.063		
Gravity tram operators	.172		
Crushermen	.185		
Blacksmiths	.109		
Superintendence	<u>.132</u>		
Total	1.713		
Average tons per man shift mining	4.67		
Milling			
Furnace operators	0.199		
Slag trammers	.199		
Superintendence	<u>.088</u>		
Total	.486		
Surface			
General labor	<u>0.199</u>		
Total	.199		

B Power and supplies

Power (k.w.h. per ton)			
Mining	11.69		
Milling	6.12		
Surface	<u>.64</u>		
Total	18.45		
Explosives (pounds per ton)			
Gelatine dynamite 35 per cent strength. Size 1 1/8 by 8		1.16	
Oil for furnaces (gallons per ton)	10.93		
Mining labor percentage of total cost		36.9	
Milling labor percentage of total cost		13.4	
Surface labor percentage of total cost		3.8	
Mining power and supplies percentage of total cost		11.3	
Milling power and supplies percentage of total cost		32.4	
Surface power and supplies percentage of total cost		<u>2.2</u>	
Total		100.0	

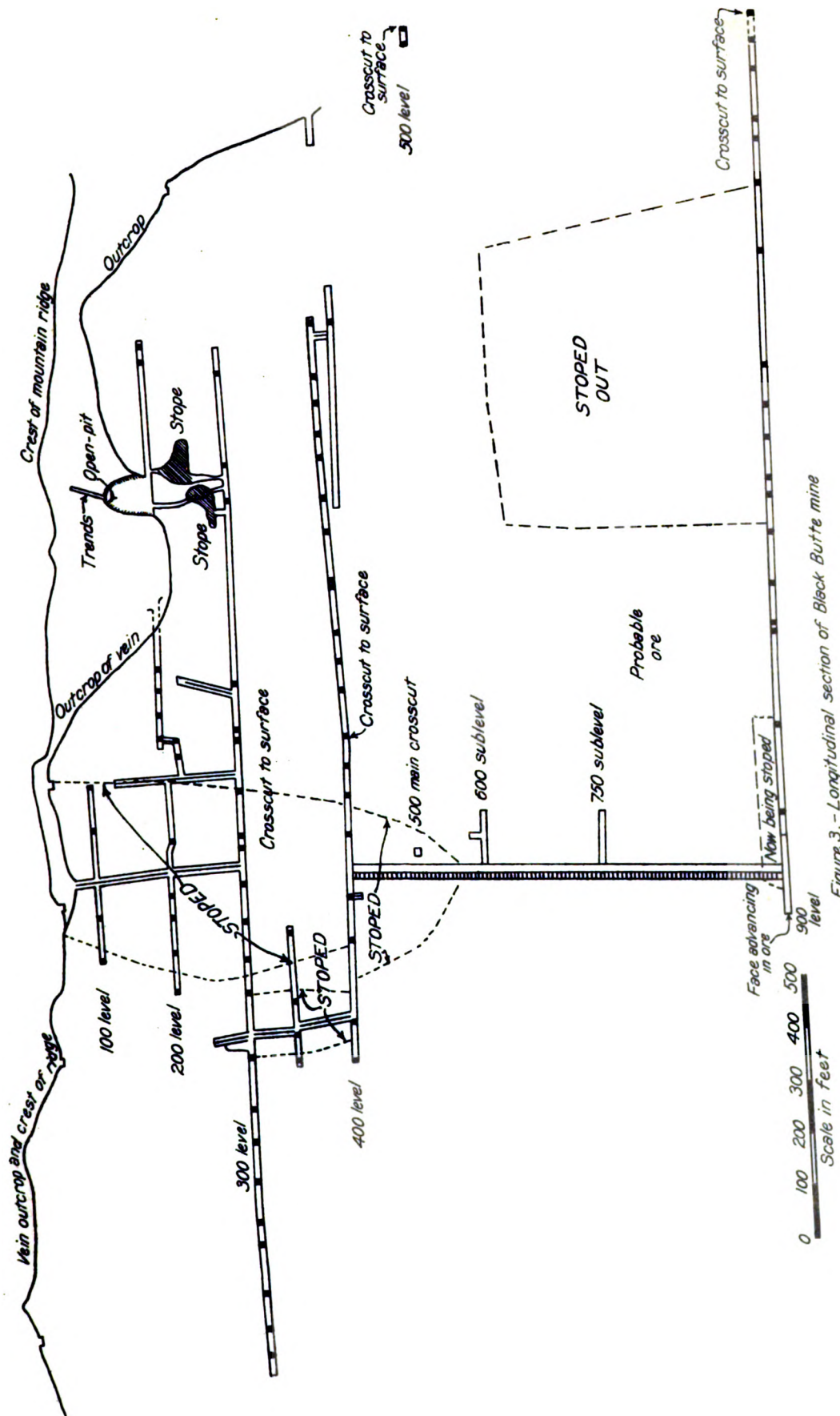


Figure 3. - Longitudinal section of Black Butte mine